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FLOWSHEET DESIGN FOR THE RECOVERY OF THE GOLD AND SILVER FROM THE ROYAL CANADIAN MINT SWEEPS

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by

T.F. BERRY*

SUMMARY OF RESULTS

Table 1 contains a head sample assay of the sweep material as received, based on the Mint assay of the crusher and screen plant metallics combined with the calculated head assays from the four tests. These results gave 277.61 oz Au/ton and 1642.5 oz Ag/ton. This method of calculation was used because, as may be seen in Tables 2 and 3, the head assays of the - 8 mesh test material resulted in a wide scatter of results attributed to a severe gold nugget effect during sampling.

Figure 1 shows a proposed crushing and screening plant flowsheet for the recovery of the coarse gold and silver metallics. This flowsheet, installed with laboratory-sized, rather than plant-sized equipment, will produce a clean high-grade gold and silver metallic concentrate. Such a concentrate, shown in Table 1, assayed 10,310.50 oz Au/ton and 17,179.4 oz Ag/ton. Based on the calculated assay of the sweep feed to the crushing plant, this metallic concentrate contained 22.3% and 6.3% of the gold and silver respectively.

Two wet and two dry concentration tests were done. These results, which are shown in the Appendix, indicate that dry concentration is to be preferred over wet concentration.

While the same approximate concentrate grade was obtained in both systems, the wet process was considerably more complex than the dry, (see Figures 2 and 3). In the wet process the use of pumps and pulp lines, which will tend to plug with coarse material, is a disadvantage. While the water used can be recycled from a head tank to the grinding, screening and tabling operations, there will be a soluble loss build-up of metal ions which will eventually have to be stripped in a water purification plant prior to

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discarding. The concentrates and tailings will have to be filtered and perhaps dried prior to refining and/or shipment to a smelter. All of these processes are added complications which are absent in a dry concentration process.

In Test 3 shown in the appendix wet concentration yielded a concentrate assaying 9260.14 oz Au/ton and 13731.7 oz Ag/ton with a recovery of 81.2% and 18.7% of the gold and silver respectively. Similarly Test 4 produced a final concentrate assaying 8266.90 oz Au/ton and 12166.0 oz Ag/ton with a recovery of 83.0% of the gold and 19.2% of the silver.

The low silver recoveries which are evident in the wet and dry concentration tests are attributable to the removal of most of the metallic silver in the crushing and screening plant. The remaining silver, largely present as compounds of silver such as AgCl, is not ammenable to recovery by gravity concentration.

The first dry concentration test (Test 1) was of a preliminary nature and was used to define the air table parameters for an optimum concentrate grade and recovery. This information was applied to the more definitive work shown in Test 2.

In this latter test, the results of which are shown in the appendix, a final concentrate assaying 9987.38 oz Au/ton and 11028.9 oz Ag/ton was produced with a recovery of 82.8% and 15.2% of the gold and silver respectively. The recleaner tailings contained 10.6% of the gold and 27.0% of the silver. It is proposed that this fraction (7.7% of the weight) form a circulating load to the cleaner circuit. Experience indicates that approximately 50% of this gold and silver would be recovered in the final concentrate. Such a procedure would give an overall recovery of 88.1% of the gold and 28.7% of the silver.

The results shown in Test 2 were achieved with one cleaner and one recleaner stage. Additional recleaner stages in a plant would reduce the final concentrate tonnage to well below the 2% by weight shown. This would result in an enhanced concentrate grade without seriously affecting the overall recovery of the gold and silver.

The combined cleaner and rougher tailings assaying 19.5 oz Au/ton and 1032.0 oz Ag/ton, containing 6.6% and 57.8% of the gold and silver respectively would be shipped to a smelter for additional recovery.
If one assumes that the calculated head assay of the sweeps shown in Table 1 is reasonably accurate, then on the basis of Test 2 the overall recovery of the gold and silver from the Mint sweeps as received would be as follows:

PRODUCT	RECOVERY %	
	Au	Ag
+6 mesh metallics	22.3	6.3
From Test 2*	68.4	26.8
Total	90.7	33.1

* 88.1% of 77.7 of the gold and 28.7% of 93.7 of the silver

The overall final concentrate grade that may be expected from the dry treatment of the Mint sweeps as outlined in Test 2, with the crushing and screening plant metallics added, can be approximated as in excess of 10,000 oz Au/ton and 12,000 Ag/ton.

ÉLABORATION D'UN SCHÉMA DE FONCTIONNEMENT POUR LA RÉCUPÉRATION DU MINERAI D'OR ET D'ARGENT CONTENU DANS LES BALAYURES DE LA MONNAIE ROYALE CANADIENNE

Le Tableau 1 comprend les résultats de l'échantillonnage des balayures qui a été effectué à l'installation de broyage et de criblage de minerai métallique de la Monnaie Royale Canadienne et les données des analyses chimiques obtenues à la suite de quatre autres essais. Les résultats indiquent que les échantillons titraient 277,61 on Au/tonne et 1 642,5 on Ag/tonne. On a utilisé cette méthode de calcul parce que, ainsi qu'il apparaît aux tableaux 2 et 3, les analyses chimiques des matériaux d'essai passant - 8 mailles causent le dispersement des résultats en raison de l'effet "pépite d'or" qui se produit pendant l'échantillonnage.

Une description du schéma de fonctionnement proposé pour l'installation de broyage et de criblage en vue de récupérer le minerai métallique d'or et d'argent est présentée au Tableau 1. Le schéma de fonctionnement élaboré au moyen d'un appareillage susceptible d'être utilisé dans un laboratoire plutôt que dans une usine permettra d'obtenir un concentré métallique à haute teneur en or et en argent. Comme l'indique le Tableau 1, ce concentré aura une teneur de 10 310,50 on Au/tonne et de 17 179,4 on Ag/tonne. Le calcul des résultats de l'échantillonnage des balayures effectué à la suite du broyage indique que le concentré métallique titrait 22,3 % d'or et 6,3 % d'argent, respectivement.

Deux essais par voie humide et deux essais par voie sèche ont été effectués. Les résultats présentés à l'annexe démontrent que la technique de concentration par voie sèche est préférable à la technique de concentration par voie humide.

Bien qu'un concentré de qualité identique ait été obtenu par les deux méthodes de traitement, le procédé par voie humide était considérablement plus complexe que le procédé par voie sèche (voir les figures 2 et 3). Dans le procédé par voie humide, l'utilisation de tuyaux et de collecteurs d'exhaure dans lesquels les matières grossières ont tendance à s'engorger, présente des inconvénients. Bien qu'il soit possible de recycler l'eau du réservoir principal vers le circuit de broyage, de filtration et de concentration, la perte d'ions métalliques sous forme soluble, lesquels devront éventuellement être extraits dans une installation d'épuration d'eau avant le

rejet, s'accroît. Les concentrés et les résidus devront être filtrés et peut-être séchés avant d'être raffinés et expédiés vers une autre fonderie. La nécessité d'utiliser tous ces procédés s'ajoute aux autres problèmes qu'il est possible d'éviter par l'utilisation du procédé de concentration par voie sèche.

Les résultats des essais de concentration par voie humide qui sont présentés à l'annexe comprennent ceux de l'essai 3 qui a permis d'obtenir un concentré titrant 9 260,14 on Au/tonne et 13 731,7 on Ag/tonne à un taux de récupération de 81,2 % et de 18,7 % pour l'or et pour l'argent, respectivement. De la même façon, l'essai 4 a donné un produit final titrant 8 266,90 on/tonne Au et 12 166,0 on Ag/tonne à un taux de récupération de 83,0 % pour l'or et de 19,2 % pour l'argent.

Les faibles taux de récupération pour l'argent qui sont indiqués de façon précise par les essais de concentration par voie humide et par voie sèche sont attribuables au fait que presque tout l'argent métallique est extrait à l'usine de broyage et de criblage. L'argent restant, qui est présent principalement sous forme de composés d'argent tels AgCl, ne se prête pas à la récupération par le procédé de concentration gravimétrique.

Le premier essai de concentration par voie sèche (essai 1), qui était de nature expérimentale, a été utilisé pour déterminer les paramètres de la table de concentration pneumatique en vue d'obtenir un taux de récupération optimal et un concentré de qualité supérieure. Ces données ont été appliquées au cours d'un deuxième essai de nature plus définitive.

Les résultats de ce dernier essai, qui sont présentés à l'annexe, indiquent que les concentrés titraient 9 987,38 on Au/tonne et 11 028,9 on Ag/tonne et que le taux de récupération était de 82,8 % et de 15,2 % pour l'argent, respectivement. Les résidus provenant de la finisseuse titraient 10,6% d'or et 27,0 % d'argent. On suggère que ces résidus (7,7 %) du poids total) soient acheminés vers le circuit de nettoyage. L'expérience démontre qu'il serait possible de récupérer environ 50 % de l'or et de l'argent contenus dans le concentré final. En procédant de cette façon, le taux de récupération global serait de 88,1 % pour l'or et de 28,7 % pour l'argent.

Les résultats du deuxième essai ont été obtenus après que les concentrés aient passé par une étape d'épuration et de refinition. Des étapes additionnelles de refinition en usine seraient nécessaires pour réduire le

tonnage final du concentré à un niveau bien inférieur au niveau indiqué qui est de 2 % (par poids). Ce processus permettrait d'obtenir un concentré amélioré sans toutefois modifier le mode de récupération pour l'or et pour l'argent.

Les résidus de dégrossissage et d'épuration titrant 19,5 on Au/tonne et 1 032,0 on Ag/tonne et contenant 6,6 % et 57,8 % d'or et d'argent, respectivement, seront expédiés à une fonderie à des fins de récupération additionnelles. En supposant que les calculs relatifs à l'échantillonnage de balayures, lesquels sont présentés au Tableau 1, soient raisonnablement exacts et en s'appuyant sur le taux de récupération global tel qu'indiqué par l'essai 2, les taux de récupération pour l'or et pour l'argent obtenus à partir des balayures de la Monnaie Royale Canadienne seraient les suivants:

PRODUIT	TAUX DE RÉCUPÉRATION	
	Au	Ag
+ 6 mailles métalliques	22,3	6,3
provenant de l'essai 2*	68,4	26,8
Total	90,7	33,1

* 88,1 % de 77,7 de l'or et 28,7 % de 93,7 de l'argent

Comme l'indique l'essai 2, le procédé de concentration par voie humide combiné aux procédés de broyage et de criblage permettrait d'obtenir un concentré titrant plus de 10 000 on Au par tonne et plus de 12 000 on AG par tonne.

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INTRODUCTION

The Royal Canadian Mint produces 1 to 2 short tons a week of a material, known as "sweeps", as a byproduct of their refinery processes. This material is composed of fire clay crucibles and cupels, slag, stems, silica-carbide pots and cottrell dust. It is an extremely high grade gold and silver byproduct.

This report contains the results of the final phase of a three phase cost recovery investigation begun in September 1985. It concerns the development of dry and wet gravity concentration processes for the recovery of the gold and silver from the sweeps in a high-grade concentrate.

The results obtained in phases I and II of the investigation are contained in Mineral Sciences Laboratories Division Report MSL 86-5 (CR).

Scope of Investigation

1) Crushing and screening and the recovery of the coarse metallics.
2) Blending and sample preparation.
3) Head sample assaying for gold and silver.
4) Grinding and screening.
5) Two dry concentration tests using an air table.
   Two wet concentration tests using a Deister laboratory table.
6) One test using an air elutriator.

Ore Shipment

On June 13, 1986 a sealed drum of sweeps was received at CANMET. A cursory examination of this material, which weighed 270 kg, indicated that it was all minus ½ inch, contained a high percentage of graphitic mineral and visible metallic gold and silver.

DISCUSSION AND EVALUATION OF TEST PROCEDURES AND RESULTS

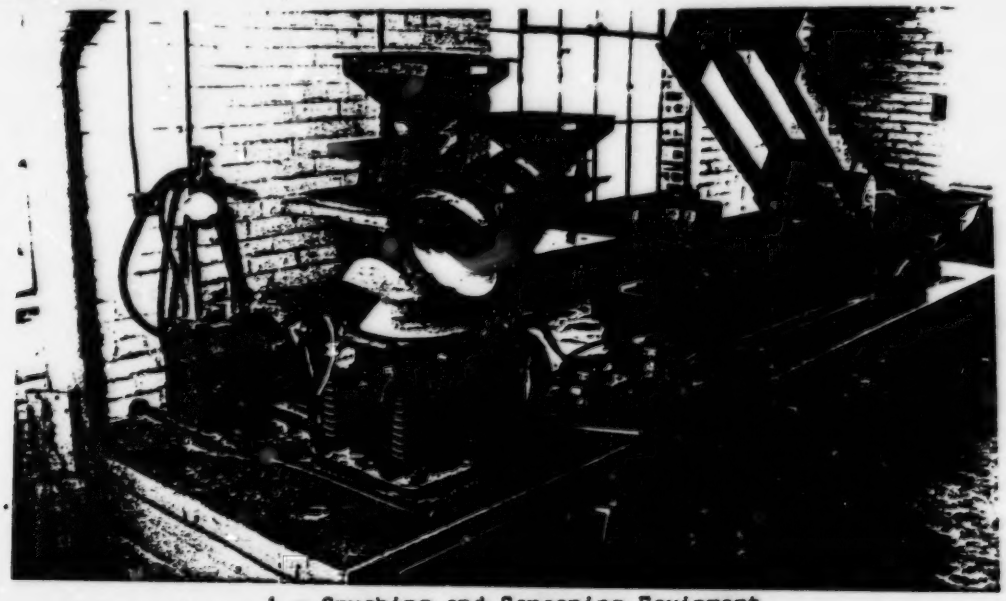
Crushing and Screening

In a semi-batch operation the contents of the drum were screened on a gently vibrating 8 mesh screen after which the +8 mesh oversize was crushed in a 12 inch Peacock cone crusher. This process was continued until all of the sweep material was reduced to -8 mesh. The gold and silver metallics were flattened during the crushing process and were removed by hand

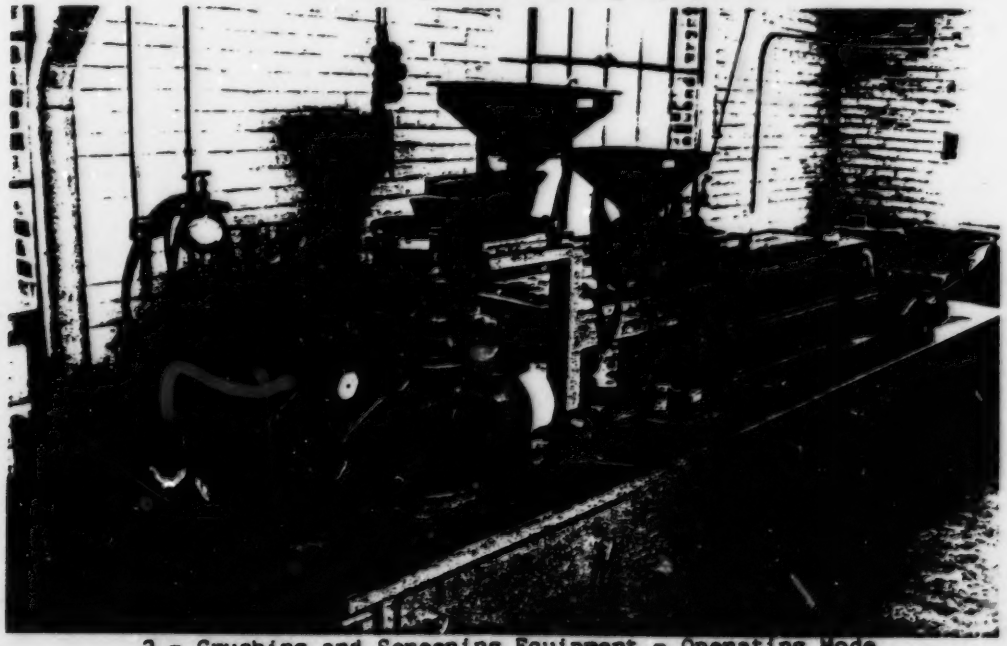
screening on a 6 mesh screen.

These metallics which were clean and very high grade were returned to the Mint for assaying. The results of this assay may be seen in Table 1.

Air, sufficient only to remove the finest dust, was applied during this crushing and screening process. The loss to a dust collector was 0.4% of the sweeps.



1 - Crushing and Screening Equipment

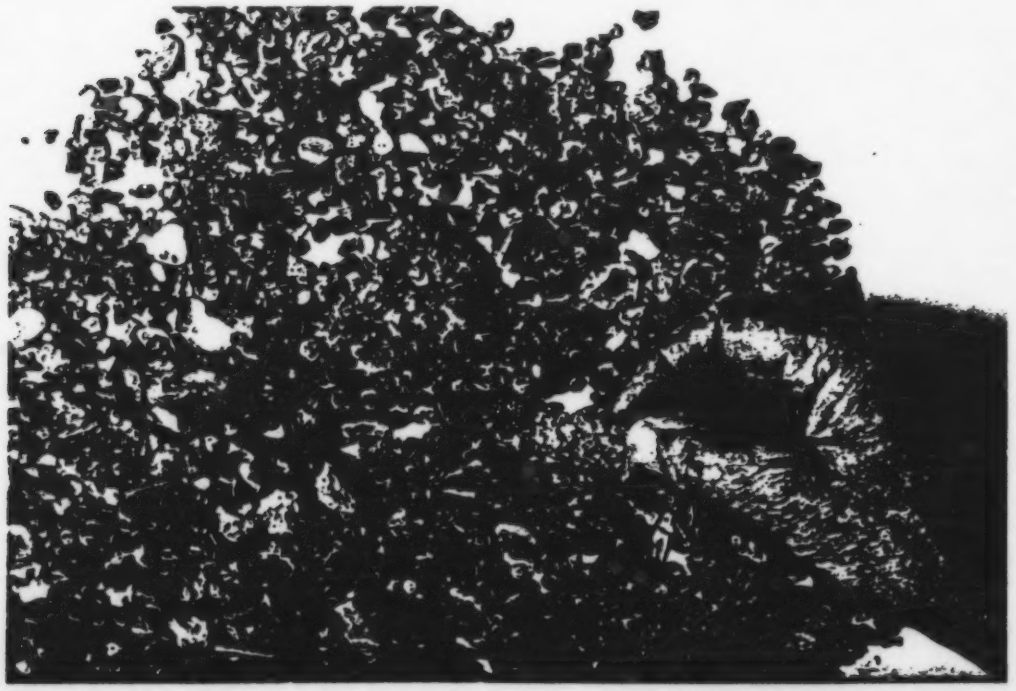


2 - Crushing and Screening Equipment - Operating Mode

It is essential to the overall recovery of the gold and silver that as large a percentage as possible of clean metallics be removed in this relatively simple and inexpensive crushing and screening operation.

Figure 1 diagrams a proposed crushing and screening plant. The sweep material is fed to a primary cone crusher set to deliver a minus 3/8 inch product. This crusher discharge falls to a vibrating 1/2 inch screen from which the coarse gold and silver metallics are removed. The screen undersize is contained in a surge bin from which it is fed to a 12 inch Peacock cone crusher set to deliver a -10 mesh product which drops onto a vibrating 6 mesh screen. The -6 mesh undersize falls into a 2 ton surge bin from which it is fed to the concentration plant.

The secondary crushing and screening should be run as a semi-batch operation with the feed to the 12 inch cone crusher being controlled to reduce as much as possible the +6 mesh circulating load. In spite of these controls the +6 mesh oversize may require additional crushing to remove unwanted gangue or until a visual examination indicates that the metallics on this screen are of sufficiently high grade to warrant their being added to the coarse primary metallics for direct refining.



3 - Metallics from Crushing and Screening Plant

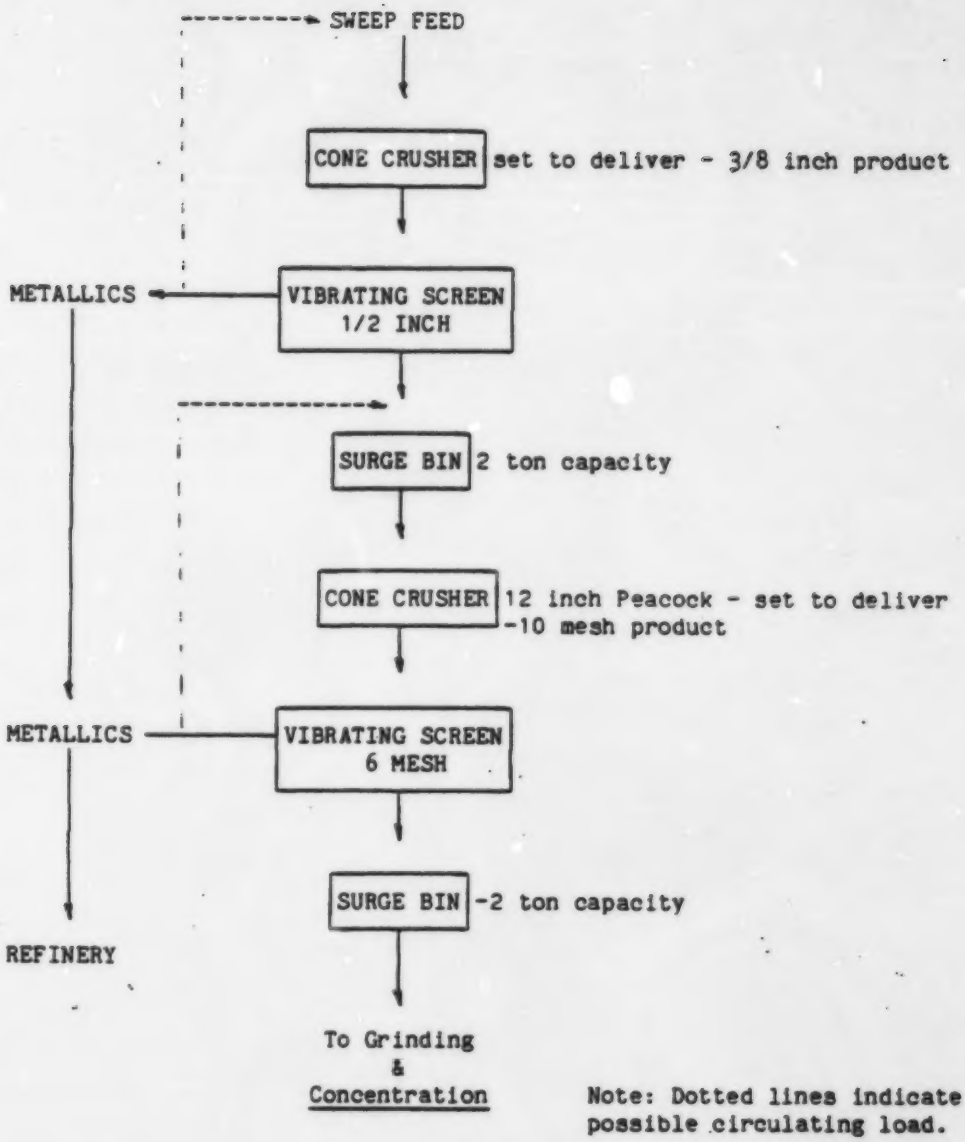


Figure 1 - Primary and Secondary Crushing and Screening

Table 1 - A Calculated Head Sample Assay

Product	Weight		oz/ton		Distribution %	
	kg	%	Au	Ag	Au	Ag
+6 mesh metallics (1)	1.66	0.6	10310.50	17179.4	22.3	6.3
-8 mesh fines (2)	267.24	99.0	217.93	1555.0	77.7	93.7
loss (calc'd) (3)	1.10	0.4	-	-	-	-
Total of sweeps (calc'd)	270.00	100.0	277.61	1642.5	100.0	100.0

(1) Mint assay.

(2) Calculated head from tests 1, 2, 3 and 4.

(3) Mechanical and dust loss. This material was lost in the CANMET crushing and screening operation and thus was not assayed. In a mill operation this material would be retained in a dust collector and would be weighed and assayed.

Blending and Sampling

The -8 mesh ore was blended in two parts in a baffled, sealed, tumbling drum for 1 hour. The blended material was removed from the drum carefully (to minimize dust losses), and was fed via vibrating hopper and launder to a pro-splitter.

4 - Pro-splitter sampler

The pro-splitter produced 20 samples which were bagged and sealed. These samples comprised the feed material for the concentration tests.

One of these samples, arbitrarily selected, was further reduced on the pro-splitter and a sample of this split was riffled into fractions for a head sample assay.

Head Sample Assay

Five head samples were prepared for assaying using a micro-splitter and a pulverizer. The results which are shown in Table 2 clearly indicate a severe gold nugget effect. This was not as apparent in the silver assays. Most of the metallic silver was removed on the crushing plant screens and that remaining was present as various compounds of silver. Sampling of any coarse sand (-8 mesh) containing a large amount of free gold will result in widely scattered gold assays.

Table 2 - Head Sample No. 1

Sample g	wt %	Assays Au	oz/ton Ag
32.3	20.2	126.16	1563.2
32.6	20.4	634.67	1529.3
32.4	20.3	432.42	1448.0
30.6	19.1	121.92	1521.0
32.0	20.0	238.92	1516.0
159.9	100.0	313.81	1524.4

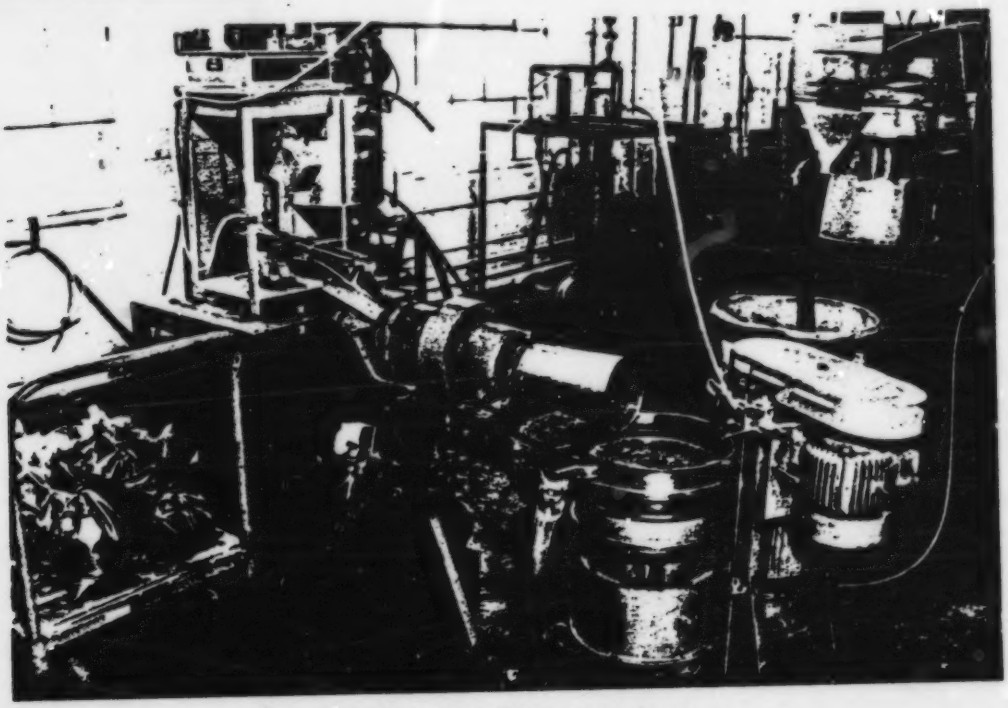
The -8 mesh sweep material was re-blended for four hours and reprocessed on the pro-splitter. Three new samples were obtained for a head sample assay. These results are shown in Table 3.

Table 3 - Head Sample No. 2

Sample g	wt %	Assays Au	oz/ton Ag
37.6	41.0	186.11	1476.8
24.0	26.2	114.37	1493.6
30.0	32.8	162.48	1493.4
91.6	100.0	159.56	1486.32

These assays, insofar as the gold is concerned, are so widely scattered that the head sample assay is meaningless. A more meaningful figure will be obtained from the calculated head during the test work. In these tests 20 to 28 kg were used. Such large samples will be more representative of the -8 mesh sweep material.

Grinding and Screening



5 - Grinding and Screening Plant

At CANMET there was no equipment available for continuous dry rod milling and screening at the low feed rates used in the four tests. All grinding and screening was done wet. Any soluble loss resulting from this wet processing would not affect the concentration characteristics on either the air or wet table.

The -8 mesh sweep material was fed at a constant rate using a vibratory feeder to a 7"x14" Denver rod mill grinding at 65% solids. The ground pulp was washed with a hard spray on a 50 mesh vibrating screen. The +50 mesh (oversize) was retained in a tank and the -50 mesh pulp was pumped to a 150

mesh screen where it was washed with a second spray. Three screen fractions were thus produced, +50, -50+150 and -150 mesh. A fourth fraction was produced by washing the +50 mesh fraction through a 20 mesh screen. The +20 mesh was a metallic concentrate.

This grinding and screening procedure was followed in each of the four tests. In Test 1 the grinding was done in open circuit and in Test 2, in a modified closed circuit. In this latter test the +50 mesh (oversize) was fed back to the rod mill by hand at approximately the same feed rate as was the original feed. Similarly, Test 3 was ground in open circuit and Test 4 in the modified closed circuit. This was an attempt to determine whether a slight regrind of the coarse fraction would increase the liberation and thus enhance the recovery of the gold and silver without seriously increasing the amount reporting in the finer fractions.

It is essential to the recovery of the gold and silver that the grinding, whether wet or dry, be closely controlled as to mill size, rod charge, feed rate and density (in the case of wet grinding) to prevent overgrinding since too fine a feed to an air table will result in unacceptable losses to a dust collector. These four tests may have erred slightly on the coarse side in an attempt to achieve optimum liberation with a minimum of fines.

Figure 2 diagrams a proposed dry grinding and concentrating plant. Here, the -6 mesh material from the crushing plant is fed to an end or centre peripheral discharge rod mill which is favoured over the ball mill to minimize over-grinding and to flatten the gold and silver metallics. The grinding parameters should be such that the gangue is preferentially ground with the discharge to a 20 mesh vibrating screen resulting in a high grade +20 mesh metallic concentrate. The -20 mesh material is screened on a 35 mesh screen with the +35 mesh oversize being returned to the rod mill. The 35 mesh screen is a compromise between the 20 and 50 mesh screens used in the tests and should ensure optimum liberation with a minimum of additional fines. The dry, -35 mesh material can now be split into two fractions using an air classifier. A 20 inch whirlwind separator of the type manufactured by Sturtevant is recommended to produce a -35+80 mesh and a -80 mesh product.

Figure 3 outlines a proposed wet grinding, screen separation and concentrating plant. The -6 mesh sweep from the crushing plant is fed to a

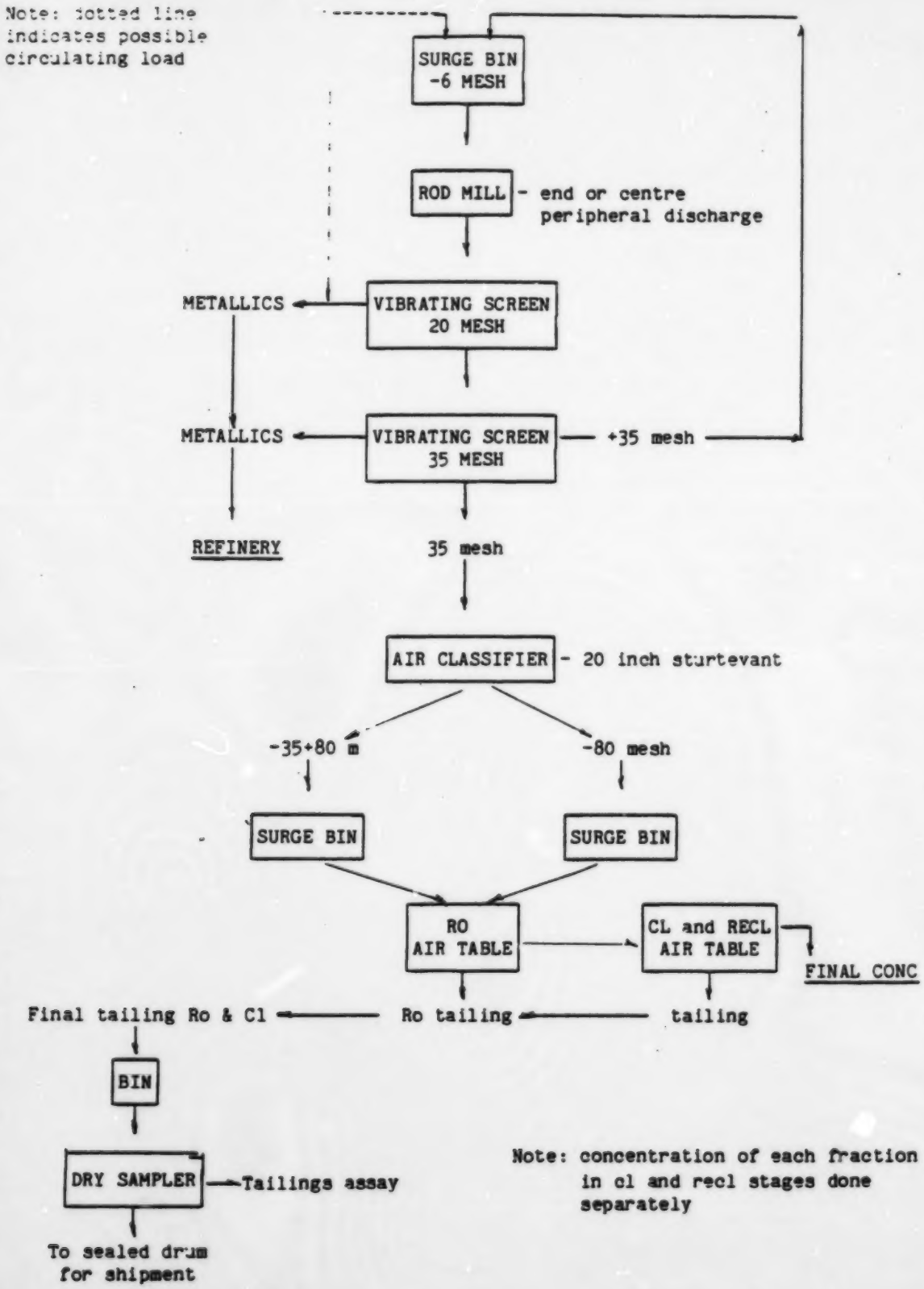
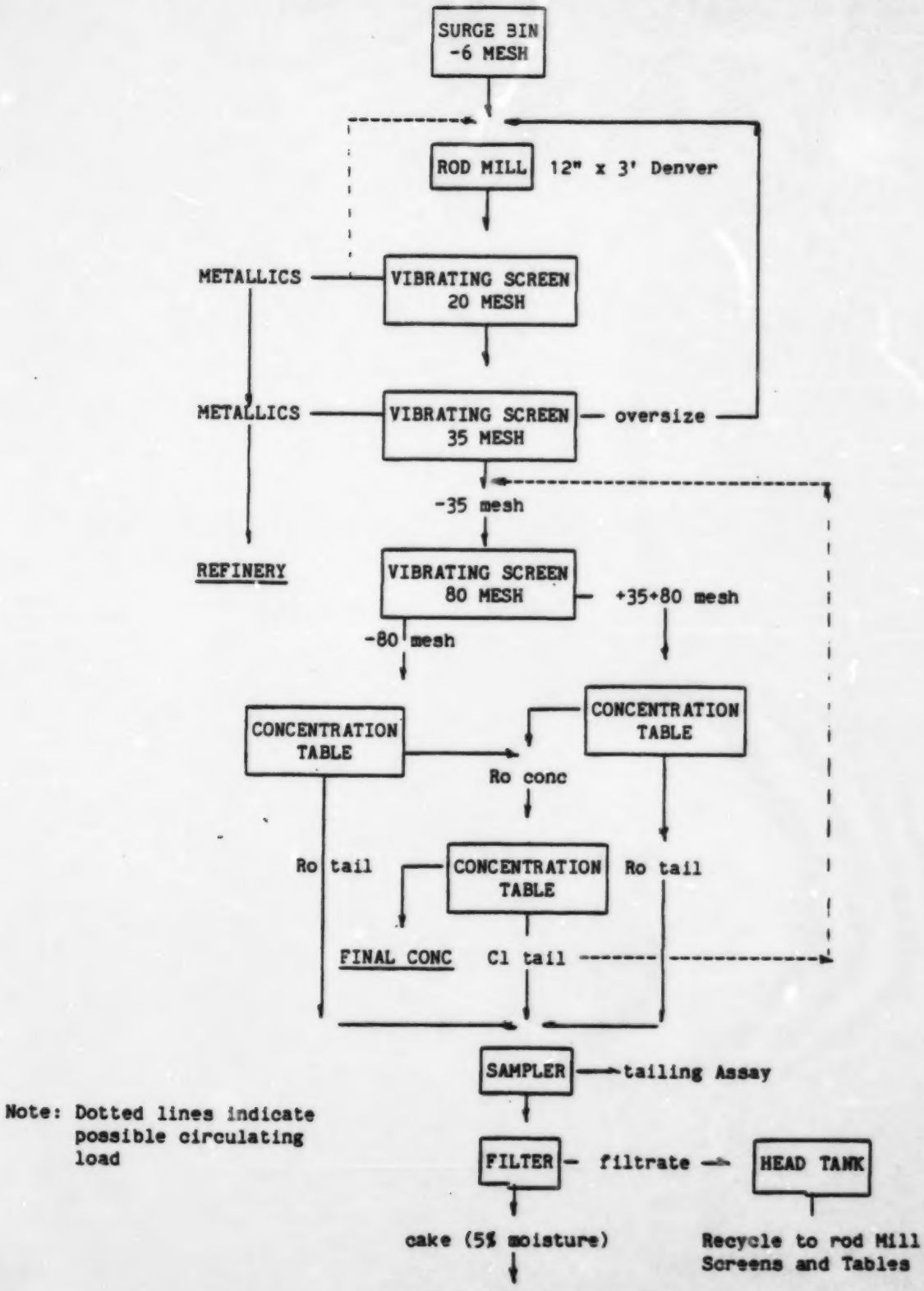


Figure 2 - Dry Grinding and Concentration



Note: Dotted lines indicate possible circulating load

Figure 3 - Wet Grinding and Concentration

12"x3' Denver rod mill grinding at 65% solids discharging to a 20 mesh vibrating screen serviced with a hard water spray from which a metallic concentrate is removed. The -20 mesh pulp is now screened on a 35 mesh vibrating screen with the +35 mesh material returning to the rod mill for additional liberation. The -35 mesh pulp is pumped to an 80 mesh vibrating screen washed with spray water producing a -35+80 mesh and a -80 mesh fraction which flow directly to the concentrating tables.

Dry Concentration (Air Table)

The air table is a shaped deck covered with an air permeable cloth through which air is blown to keep the particles on the deck in motion. The horizontal pitch of the table and the lateral slope can be adjusted to suit the feed particle size. The air can be controlled, as can the motion of the table. The air exhausted above the table is drawn off to a dust collector.

The KU-1095 air table was used in the tests. Another larger but still laboratory sized table was too large for the amount of sweep material under test.

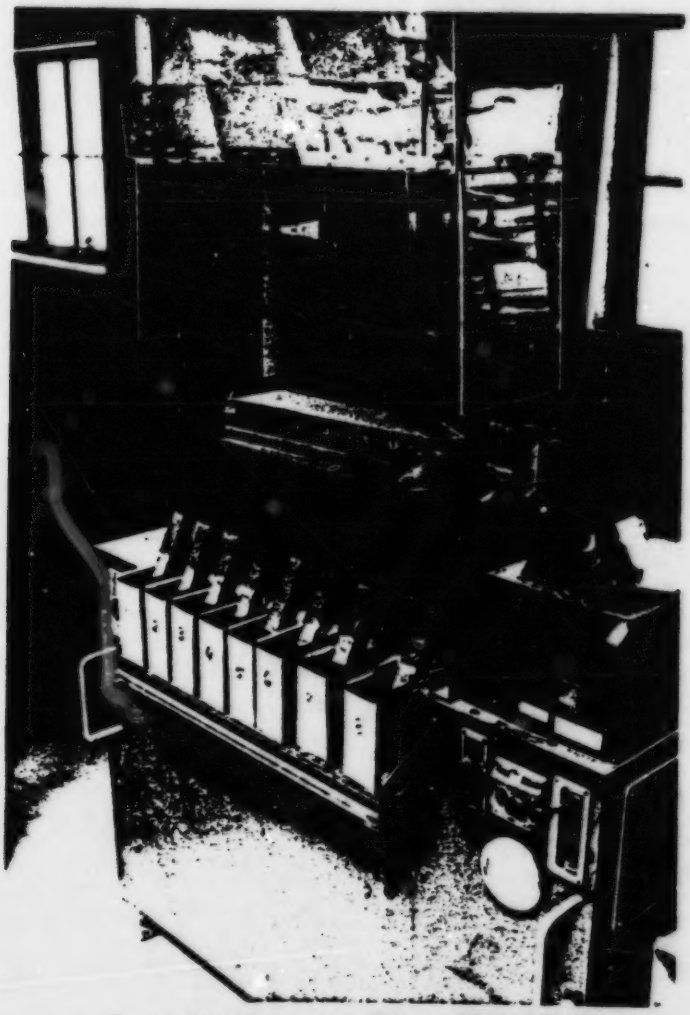
The results of Test 1 and 2 shown in the appendix were obtained using a binocular microscope as a contral on the concentrate grade. This is the simplest and fastest means available for setting the table parameters.

In Test 1, the +20 mesh metallics, -20+50 mesh and the -50+150 mesh fractions were concentrated on the air table. Because of unacceptable dust losses on this table, the -150 mesh fraction was concentrated on a wet table.

This preliminary dry concentration test was designed to determine the air table conditions necessary for maximum concentrate grades and recoveries. The knowledge gained was applied to the more definitive Test 2.

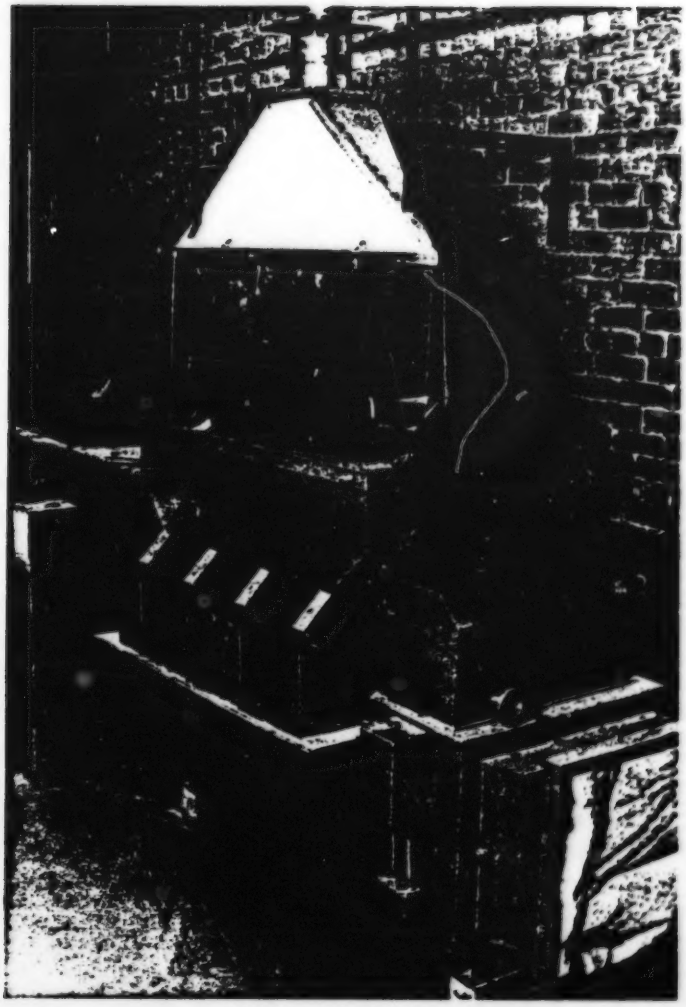
The combined final concentrate assayed a calculated 7382.39 oz Au/ton and 9996.1 oz Ag/ton with a gold recovery of 79.6%. The silver did not concentrate. The metallic silver was largely removed in the crushing section and the remaining silver was present as silver compounds such as silver chloride. This light, fine material reported in the tailing fractions. The combined rougher tailings assayed a calculated 25.94 oz Au/ton and contained 10.1% of the gold. The addition to these tailings of all cleaner tailing fractions resulted in a final disposable tailing

assaying 39.41 oz Au/ton and 1588.4 oz Ag/ton. This final tailing contained 20.4% and 88.5% of the gold and silver respectively in the original feed to concentration.



6 - Laboratory Air Table (KU-1095)

In the dry concentration Test 2 a final concentrate assaying 9987.38 oz Au/ton and 11028.9 oz Ag/ton was produced with a recovery of 82.8% of the gold and 15.2% of the silver. As in all of the tests the silver was not ammenable to gravity concentration. The combined cleaner and rougher tails assayed 19.5 oz Au/ton and contained 6.6% of the gold..

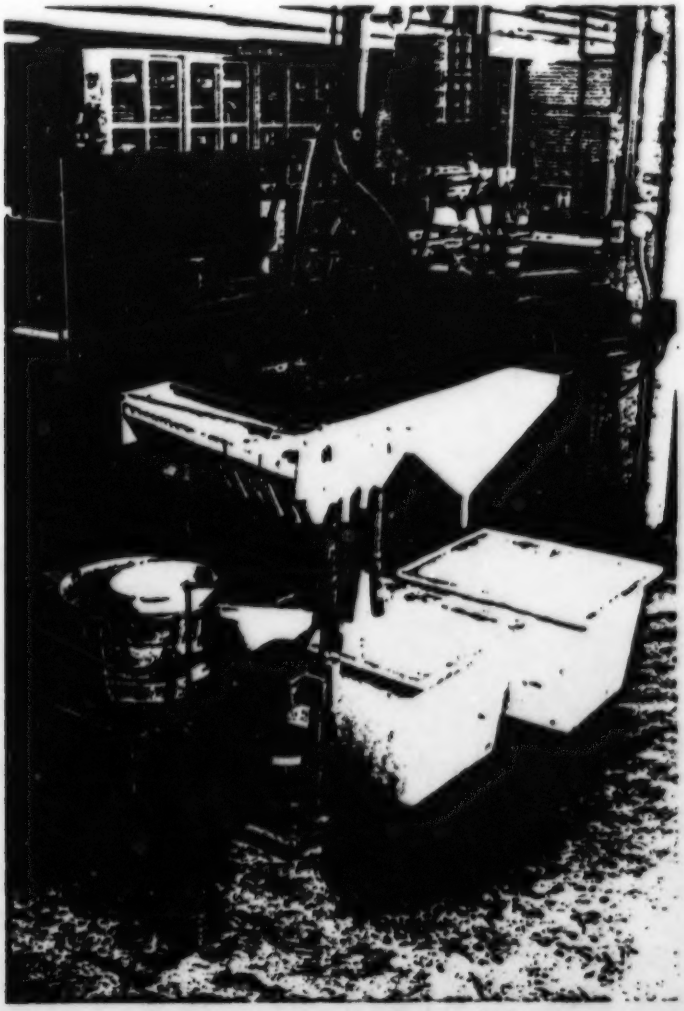


7 - Laboratory Air Table (10C-No2 Deck)

As was the case in Test 1 the -150 mesh fraction was concentrated on a wet table. The rougher concentrate was cleaned once. The cleaner tail assaying 937.28 oz Au/ton had been added to the recleaner tails from the other two fractions and assays a combined 232.83 oz Au/ton and contains 10.6% of the gold. This fraction amounting to 7.7% of the weight should form a circulating load to the air table. At least 50% of this gold would be recovered in the final concentrate.

As mentioned, the -150 mesh fractions in Tests 1 and 2 were concentrated on a wet table. This was done to avoid the high dust losses which would result if an air table had been used. In a properly designed plant the concentration of this fine gold on an air table should yield results comparable to those shown in the Appendix in Test 2. The exhausted dust, which would be high grade, would be contained in a suitable dust collector.

Wet Concentration (Deister Table)



8 - Deister Laboratory Concentrating Table

Wet concentration was done using a Deister laboratory table fitted with a sand deck.

In each of the two wet concentration tests a +20 mesh metallic concentrate was removed and the three remaining screen fractions (-20+50, -50+150 and -150 mesh) were concentrated separately. The rougher concentrates which were obtained were cleaned once on the table.

In Test 3, a +20 mesh metallic concentrate in 0.7% of the weight assayed 16,192.00 oz Au/ton and 10,879.6 oz Ag/ton. Similarly in Test 4 in 1.0% by weight the +20 mesh metallics assayed 12,778.70 oz Au/ton and 7,461.4 oz Ag/ton. In each of these metallic concentrates approximately 50% of the gold in the feed was recoverd.

In Test 3 and 4, the -20+50 mesh rougher tailing assayed 47.01 and 65.10 oz Au/ton respectively. These, higher than expected assays, representing 10.2% and 9.4% of the gold may be the result of a lack of liberation. In Figures 2 and 3 the indicated grind prior to concentration (wet or dry) is -35 mesh rather than the -20 mesh used in the test work. This slight increase in grinding should result in an increased gold recovery without seriously increasing the amount of fines produced.

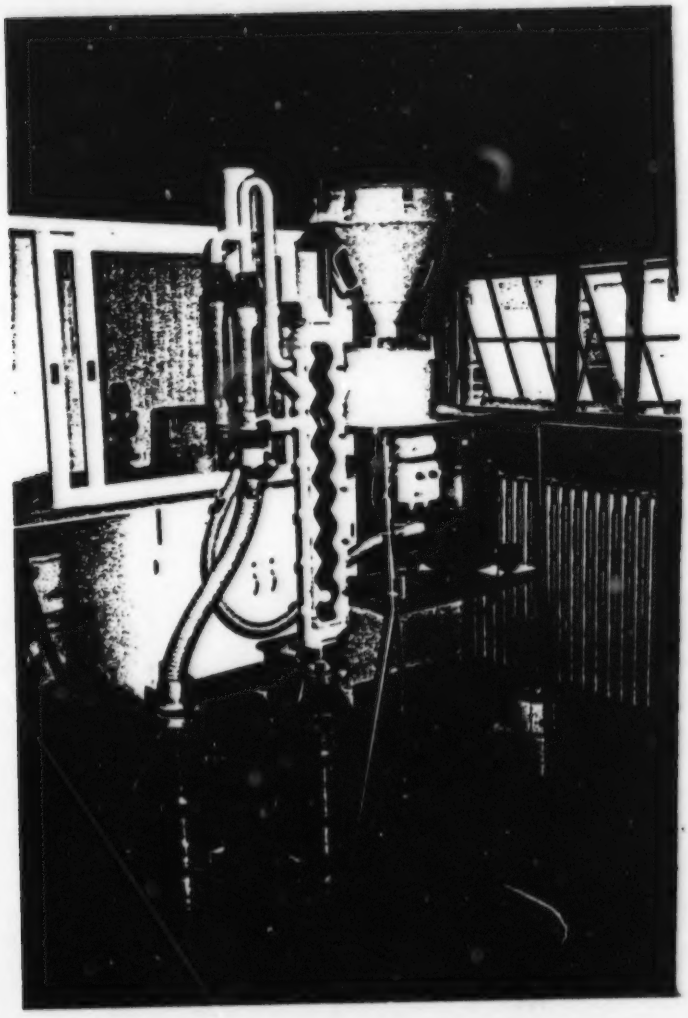
Dry Concentration (Zig-Zag Classifier)

A small portion of each of the screened fractions produced in Test 2 was removed for an experimental test on this equipment. This concentration device is simply an air elutriator. A metered flow of air is swept up the chamber against the falling particles removing the light material and allowing the heavier particles to concentrate in a glass container. A more closer screen sizing of the feed to this equipment is essential than is the case with an air table.

The light fraction swept out of the chamber is cycloned with the coarse middling being retained in a second glass container and the light (dust) being trapped on a filter paper.

This test was an attempt to find an alternative to air table concentration.

A microscopic examination of the product showed no improvement over air table concentration and the products were not assayed.



9 - Zig-Zag Classifier

CONCLUSIONS

It is difficult to project a flowsheet on the basis of the results of two tests (wet or dry). Nevertheless we feel certain that the flow diagrams outlined in Figures 1 and 2, or some modification of these equipment assemblies, will return high-grade concentrates with a high recovery of the gold and silver from a dry process.

Such a process is favoured over the wet process for several reasons. In dry processing all dust losses are automatically recovered in a dust collector from hooded equipment. Likewise all mechanical losses can be recovered by sweeping and vacuuming. There is no soluble loss.

In a wet process all concentration must be filtered and dried prior to refining and the final tailings will require filtering prior to shipment of a wet cake (5% moisture) to a smelter. While the process water containing soluble metals can be impounded in a head tank from which it can be recycled to the grinding, screening and table concentration sections, it eventually will have to be stripped of its contained metals. Thus a wet process is somewhat more complex to handle than is a dry process.

Dry concentration breaks down into these distinct semi-batch operations. It is recommended that each phase of the flowsheet be completed before proceeding to the next phase.

1) Crushing with removal of coarse metallics on screens;
2) Grinding with removal of additional metallics, and production of two fractions using an air-swept separator;
3) Dry concentration of the two fractions on an air table.

Even with the primary and secondary crushers set to close tolerances.
there is bound to be some circulating load. Re-crushing the oversize one or more times may be necessary before a satisfactorily clean, high grade, +6 mesh metallic concentrate is obtained. If the work is done carefully, no problems are foreseen in this area.

Grinding should be done in a rod mill (either a centre, or an end peripheral discharge) as opposed to a ball mill. A rod mill is basically a tertiary crusher and from such equipment the product will be coarser than that obtained from a ball mill. The production of fines is thus reduced. The rod mill will flatten the metallic gold and silver ensuring an easier recovery on a concentrating table. A ball mill tends to ball the gold and silver metallics into spheres which makes their concentration on a table much more difficult.

No problems are foreseen in dry screening at the relatively coarse sizes of 20 and 35 mesh. From each of these screens a metallic concentrate will be removed. In the grinding and screening phase there will be a circulating load (perhaps small) as the sweep material is ground to -35 mesh. As mentioned previously, this screen is a compromise between the 20

and 50 mesh screens used in the tests, and should provide an increased liberation and by extension, an increased recovery, with a minimum increase in fines.

The -35 mesh material will best be separated into two fractions (± 80 mesh) using an air-swept classifier. A 20-inch Sturtevant should be sufficient for this separation with each fraction being held in its own bin for subsequent air table concentration.

As already described in the section under dry concentration the small laboratory (KU-1095) Kipp Kelley air table was used during the test work. A larger but still laboratory-sized table (100 -NO2 deck) was considered too large for the amount of sweep material under test, but should be ideal for rougher concentration in a plant. The smaller table could then be used for the cleaner and recleaner upgrading operations. Each fraction (± 80 mesh) should be concentrated separately.

The table parameters i.e. vibration control, air volume, deck pitch and slope can easily be adjusted during operation. The use of a binnocular microscope, rather than assaying, is recommended as the quickest and easiest means of adjusting the table parameters to achieve optimum concentrate grades and recoveries.

The rougher table tailings and (if low assays warrant) the cleaner tailings are consigned to a bin from which they are fed through an automatic dry sampler to drums for shipment to a smelter. Table recleaner tailings representing a small portion of the sweeps, but high grade, will form a circulating load to the tables.

It is recommended that no attempt be made to obtain a head sample assay on the -6 mesh feed to a concentrating plant. Such head assays as shown in Tables 2 and 3 are bound to be suspect due to a severe gold nugget effect. A calculated head based on the weights and assays of the concentrates and tailings will be much more meaningful.

Dust and mechanical losses in a properly run plant should be low. To weigh, sample and assay these losses in order to complete a metallurgical balance will mean the installation of separate portable dust collecting equipment.

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APPENDIX

Test 1 - Dry Concentration - Test Data (open circuit)

Feed to Hopper 28.15 kg
Feed rate to rod mill 211.9 g/min
Water - Rod mill 120 mL/min - 65% solids
50 mesh screen 220 mL/min
150 mesh screen 240 mL/min

Total water - 540 mL/min at feed rate = 2468 L/short ton dry feed.

Solution analysis mgm/L - Fe - 0.17
Cu - 1.30
Zn - 0.22

Screen Test -	wt %
Metallics	1.9
-20+50 mesh	47.6
-50+150 mesh	36.6
-150 mesh	13.9

Losses - Mechanical, dust, solution
Grind - 1.9%
Air table - 4.9%
Total - 6.8%

In this test the +20 mesh metallics were tabled separately. A rougher concentrate was produced from the -20+50 mesh fraction and was cleaned three times. A rougher concentrate from the -50+150 mesh fraction was cleaned twice. The -150 mesh fraction was wet tabled to produce a rougher concentrate which was cleaned once.

Test 1 - Fractional Metallurgical Balance

Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Distribution % Au	Distribution % Ag
+20 mesh metallics					
conc No 1	6.9	17026.50	6819.6	22.9	11.4
conc No 2	28.0	11068.00	9475.8	60.4	64.2
conc No 3	20.5	3634.00	3404.6	14.5	16.9
ro. tail	44.6	253.30	702.4	2.2	7.5
Head (Calc'd)	100.0	5131.90	4134.9	100.0	100.0
1 + 2 + 3	55.4	9059.40	6898.4	97.8	92.5
-20+50 mesh					
Final conc	1.3	5849.70	13653.0	76.3	14.3
3rd cl tail	1.3	291.96	8507.3	3.8	8.9
2nd cl tail	4.3	120.18	4425.8	5.1	15.3
1st cl tail	30.2	23.48	1118.3	7.1	27.2
ro. tail	62.9	11.97	679.9	7.7	34.3
Head (calc'd)	100.0	99.63	1243.8	100.0	100.0
-50+150 mesh					
Final conc	0.6	2521.90	10917.0	23.2	3.1
2nd cl tail	1.1	317.15	4015.0	5.3	2.1
1st cl tail	16.1	89.51	2586.9	22.1	19.6
ro. tail	82.2	39.22	1937.9	49.4	75.2
Head (calc'd)	100.0	65.27	2119.1	100.0	100.0
-150 mesh (wet)					
cleaner conc	0.4	9529.70	17190.0	43.2	3.1
cleaner tail	7.2	540.98	11002.0	44.1	36.2
ro. tail	92.4	12.14	1435.0	12.7	60.7
Head (calc'd)	100.0	88.29	2186.8	100.0	100.0

Test 1 - Consolidated Metallurgical Balance

Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Distribution % Au	Distribution % Ag
+20 m metallics 1,2,3 conc	1.0	9059.40	6898.4	51.4	4.2
-20+50 m Final conc	0.6	5848.70	13653.0	19.9	5.2
-50+150 m conc	0.2	2521.90	10917.0	2.9	1.3
-150 m conc	0.1	9529.70	17190.0	5.4	1.0
Total conc (calc'd)	1.9	7382.39	9996.1	79.6	11.5
-20+50 m cl tail	15.9	44.84	1783.9	4.0	17.3
-50+150 m cl tail	5.9	104.84	2683.7	3.5	9.7
-150 m cl tail	0.9	540.98	11002.0	2.8	6.1
Total cl tail (calc'd)	22.7	80.13	2383.2	10.3	33.1
+20 m ro. tail	0.8	253.30	702.4	1.1	0.3
-20+50 m ro. tail	27.8	11.97	679.9	1.9	11.5
-50+150 m ro. tail	28.0	39.22	1937.9	6.2	33.0
-150 m ro. tail	12.0	12.14	1435.0	0.9	10.6
Total ro. tail (calc'd)	68.6	25.94	1325.9	10.1	55.4
Total loss	6.8	-	-	-	-
Head (calc'd)	100.0	176.25	1641.6	100.0	100.0
Combined cl + ro. tails	91.3	39.41	1588.8	20.4	88.5

Test 2 - Dry Concentration - Test Data (modified closed circuit)

Feed to Hopper 19.15 kg
Feed rate to rod mill 240 g/min
Water - Rod mill 120 mL/min - 66.7% solids
50 mesh screen 220 mL/min
150 mesh screen 200 mL/min

Total water - 54.0 mL/min at feed rate = 2037 L/short ton dry feed.

Solution analysis mgm/L - Fe - 0.13
Cu - 1.49
Zn - 0.10

Screen Test -	wt %
Metallics	1.2
-20+50 mesh	48.63
-50+150 mesh	23.5
-150 mesh	27.0

Losses - Mechanical, dust, solution
Grind - 2.6%
Air table - 3.1%
Total - 5.7%

The +50 mesh fraction was hand fed to the rod mill at the same approximate feed rate and % solids as the original feed, for increased liberation. No account was made of the increased water used.

The +20 mesh metallics produced by screening the +50 mesh material on a 20 mesh screen were held for direct assaying. In Test 1 this fraction was concentrated separately on the air table.

The -150 mesh fraction was wet tabled to avoid losses to the dust collector.

Test 2 - Fractional Metallurgical Balance

Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Distribution % Au	Distribution % Ag
+20 mesh metallics	-	11569.40	8954.7	-	-
-20+50 mesh					
rec'l conc	0.9	8282.30	14025.1	64.2	10.2
rec'l tail	7.1	373.47	4383.0	20.4	25.3
cl conc (calc'd)	8.0	1375.71	5467.7	84.6	35.5
cl tail	19.6	27.33	1101.0	4.2	17.6
ro. conc (calc'd)	27.6	418.16	2366.7	88.8	53.1
ro. tail	72.4	20.19	797.8	11.2	46.4
Head (calc'd)	100.0	130.03	1230.8	100.0	100.0
-50+150 mesh					
rec'l conc	1.0	4076.46	12772.7	57.0	8.8
rec'l tail	16.0	122.97	3407.2	27.5	37.4
cl conc (calc'd)	17.0	355.53	3958.1	84.5	46.2
cl tail	27.8	23.02	1418.4	8.9	27.0
ro. conc (calc'd)	44.8	149.19	2382.1	93.4	73.2
ro. tail	55.2	8.56	707.4	6.6	26.8
Head (calc'd)	100.0	71.56	1457.7	100.0	100.0
-150 mesh (wet)					
cl conc	0.7	7816.30	17592.0	49.5	6.1
cl tail	3.9	937.28	14004.7	33.1	27.2
ro. conc (calc'd)	4.6	1984.09	14550.6	82.6	33.3
ro. tail	95.4	20.13	1407.3	17.4	66.7
Head (calc'd)	100.0	110.47	2011.9	100.0	100.0

Test 2 - Consolidated Metallurgical Balance

Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Distribution % Au	Distribution % Ag
+20 mesh metallics	1.2	11569.40	8954.7	57.5	7.3
-20+50 mesh recl conc	0.4	9282.30	14025.1	15.4	3.8
-50+150 mesh recl conc	0.2	4076.46	12772.7	3.4	1.7
-150 mesh cl conc	0.2	7816.30	17592.0	6.5	2.4
Total conc. (calc'd)	2.0	9987.38	11028.9	82.8	15.2
-20+50 mesh recl tail	3.2	373.47	4383.0	4.9	9.5
-50+150 mesh recl tail	3.5	122.97	3407.2	1.8	8.0
-150 mesh cl tail	1.0	(1)937.28	14004.7	3.9	9.5
Total recl tail (calc'd)	7.7	232.83	5189.0	10.6	27.0
-20+50 mesh cl tail	8.9	27.33	1101.0	1.0	6.6
-50+150 mesh cl tail	6.0	23.02	1418.4	0.6	5.8
Total cl tail (calc'd)	14.9	25.59	1228.8	1.6	12.4
-20+50 mesh ro. tail	32.9	20.19	797.8	2.7	17.8
-50+150 mesh ro. tail	11.9	8.56	707.4	0.4	5.7
-150 mesh ro. tail	23.0	20.13	1407.3	1.9	21.9
Total ro. tail (calc'd)	67.8	12.29	670.3	5.0	45.4
Total loss	7.6	-	-	-	-
Head (calc'd)	100.0	241.48	1477.2	100.0	100.0
Combined cl and ro tails	82.7	19.5	1032.0	6.6	57.8

(1) The -150 mesh cl tail is rich enough to be added to the recl tails. This fraction (7.7% of the wt) would comprise a circulating load to the air table.

The combined cl and ro tails (82.7% of the wt.) would be shipped to a smelter.

Test 3 - Wet Concentration - Test Data (open circuit)

Feed to Hopper 19.25 kg

Feed rate to rod mill 220 g/min - 64.7% solids

Water - Rod mill 120 mL/min

50 mesh screen 120 mL/min

150 mesh screen 200 mL/min

Total water - 54.0 mL/min at feed rate = 2179 L/short ton dry feed.

Solution analysis mgm/L - Fe - 0.13

Cu - 1.20

Zn - 0.20

Losses - Mechanical, dust, solution - 9.1%

Test 3 - Metallurgical Balance

Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Distribution % Au	Distribution % Ag
+20 mesh metallics	0.7	16192.00	10879.6	49.7	5.2
table cl conc	1.3	5527.60	15267.5	31.5	13.5
Table cl tail	5.4	229.69	4167.8	5.4	15.3
Table ro. conc (calc'd)	6.7	1257.63	6321.6	36.9	28.8
-20+50 mesh ro. tail	49.4	47.01	922.6	10.2	30.9
-50+150 mesh ro. tail	13.4	14.08	894.3	0.8	8.1
-150 mesh ro. tail	20.7	26.11	1924.3	2.4	27.0
Total loss	9.1	-	-	-	-
Head (calc'd)	100.0	228.12	1473.6	100.0	100.0
-20 mesh metallics plus table cl conc:	2.0	9260.14	13731.9	81.2	18.7
Combined ro. tailing	83.5	36.54	1166.4	13.4	66.0

Test 4 - Wet Concentration - Test Data (modified closed circuit)

Feed to Hopper 18.478 kg
Feed rate to rod mill 230.5 g/min - 65.7% solids
Water - Rod mill 120 mL/min
50 mesh screen 220 mL/min
150 mesh screen 200 mL/min

Total water - 54.0 mL/min at feed rate = 2117 L/short ton dry feed.

Solution analysis mgm/L - Fe - 0.13
Cu - 0.31
Zn - 0.20

Losses - Mechanical, dust, solution - 7.5%

Test 4 - Metallurgical Balance

Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Distribution % Au	Distribution % Ag
+20 mesh metallics	1.0	12778.70	7461.4	51.3	4.7
table cl conc	1.5	5259.04	15302.4	31.7	14.5
table cl tail	5.3	206.10	4941.7	4.4	16.6
Table ro. conc (calc'd)	6.8	1320.72	7227.1	36.1	31.1
-20+50 mesh ro. tail	35.9	65.10	861.9	9.4	19.6
-50+150 mesh ro. tail	21.7	13.30	836.8	1.2	11.5
-150 mesh ro. tail	27.1	19.00	1930.0	2.0	33.1
Total loss	7.5	-	-	-	-
Head (calc'd)	100.0	249.00	1580.1	100.0	100.0
-20 mesh metallics plus table cl conc:					
	2.5	8266.90	12166.0	83.0	19.2
Combined ro. tailing	84.7	37.08	1197.2	12.6	64.2